ATTACHMENT C

RECEIVED
JUL 30 1982
DIVISION OF
OIL, GAS & MINING

CONFIDENTIAL

Cleveland-Cliffs

Helping develop earth's resources to meet the world's needs.

WESTERN DIVISION

RIFLE, COLORADO

WHITE RIVER SHALE PROJECT

PRELIMINARY ROCK MECHANICS REPORT

OCTOBER 2, 1975

BY

DR. ROBIN N. DATTA

ABSTRACT

Rock mechanics data collected so far for the White River Shale Project is presented in this report.

Preliminary designs for the room and pillar dimensions of the projected mine based on available data indicate that an overall recovery rate of 70% is achievable. It is shown that the rate of recovery can be further improved by dividing the area to be mined into two or more regions and employing different room and pillar dimensions in each region.

Additional rock mechanics work considered necessary for refining the designs and gaining greater confidence in them is outlined.

CONTENTS

CONTENTS

LIST OF TABLES AND DIAGRAMS

TABLES

DIAGRAMS

PLATES

1. INTRODUCTION

The White River Shale Project is scheduled to mine and retort oil shale from Ua and Ub, two federal leased tracts in Uintah County, Utah. As mining consultant of this project, the Western Division of The Cleveland-Cliffs Iron Company (CCIC) is at present engaged in preparing a comprehensive mining plan. CCIC gives due recognition to the fact that considerable long term benefits can be accrued by designing a mine on the basis of sound rock mechanics principles. As such, collection of rock mechanics data has been an important aspect of the exploratory drilling program and continues to be so in subsequent phases of the project.

A total of twenty exploratory drill holes have been drilled within and around Ua and Ub by various organizations over the past four years. Core was recovered from various intervals of interest in each hole. The thirteen holes of the drilling program conducted in late 1974 and early 1975 were logged during drilling and selected portions of the core were sent to the Bureau of Mines in Denver for various tests. Various geophysical logs were run in the holes of which the 3-D velocity log and density log were specifically performed for deriving certain mechanical properties of the rocks. Data collected so far range from logs of observable structural features like joints, partings, vugs, etc., elastic and other property logs derived from geophysical logs, and results of laboratory tests on cores for various mechanical properties.

This report presents the rock mechanics data collected to date for the White River Shale Project together with the preliminary room-and-pillar design, predicted extraction ratio, support requirement forecast, etc. for the proposed mine based on this data.

Further data collection, considered necessary to gain precise knowledge about the structural features and the virgin stressfield in the area concerned, is specified.

Additional analytical work required for refining the designs and for gaining greater confidence in them is outlined.

2. RELEVANT GEOLOGY

Geological information about the deposit to be mined and the surrounding ground is a basic requirement for any mining project. As such, information regarding the oil shale deposits in Ua and Ub gathered by exploratory drilling and other means has been presented in a separate geologic report. The geological section of this report is restricted to geological information about oil shale deposits in general and those in Ua and Ub in particular that is of special interest to one designing an oil shale mine.

2.1 Stratigraphy: Laminations and Continuity of Beds.

The oil shale deposits of immediate economic potential occurring in northwestern Colorado and northeastern Utah belong to the same lithologic unit called the Parachute Creek member of the Green River formation (Eocene). This latter formation consists of lacustrine sediments of sandstones, shales and marlstones deposited in separate basins in the two areas, namely Piceance Creek basin in Colorado and Uintah basin in Utah. The Green River formation occurs in the Washakie and Green River basins in Wyoming as well. The richest and thickest sequence of oil shale beds in both the Piceance Creek basin and Uintah basin occur at some intermediate horizon in the Parachute Creek member and is called the Mahogany zone. A yet unspecified height in the Mahogany zone in the two tracts Ua and Ub is projected to be mined by the White River Shale Project.

The two characteristic features of oil shale deposits of particular interest to the mine designer are: 1) highly laminated nature of the rock and 2) continuity of beds. Visually, oil shale which is a type of marlstone containing an oil yielding organic substance called kerogen, displays a remarkably high degree of lamination. However, unlike in some metamorphic rocks such as schist, slate, etc. the laminations are not due to any preferred orientation of some flaky mineral but due to variations in prevalent conditions during different cycles of deposition. Such variations have imparted different characteristics in terms of color, physical properties, organic matter content, etc. to the various layers of deposition. The implication of this is that extrapolation of rock properties required for mine design from one bedding horizon to another is not advisable and one has to be very careful while attempting to do so. On the positive side, since the laminations are not due to the preferred orientation of flaky minerals, the physical bond between the laminae is expected to be stronger than is usually the case with schistose or slaty rocks unless specifically weakened by a physical discontinuity such as a parting plane.

The other characteristic of oil shale deposits, that is lateral continuity of beds, is as remarkable as the laminations. Distinctive oil shale beds can be identified over very long distances and this is particularly the case with several thin tuff beds which occur in the Parachute Creek member of the Green River formation. One such tuff bed averaging no more than 0.5 feet in thickness occurs in the Mahogany zone of both the Uintah and Piceance Creek basins and is called the Mahogany marker. The Mahogany marker has been extensively used in literature as a reference datum and will continue to be so used in the future. The Mahogany marker will be used as a reference marker in this report with the positive sign indicating above the marker and the negative sign indicating below it. The advantage of having continuous beds of rock is that lateral extrapolation of rock properties in a particular horizon is allowable and information from a limited number of suitably positioned locations can yield representative values for any rock property for a particular horizon over a wide area.

2.2 Descriptive Logs of Cores from Exploratory Drill Holes for the Mining Zone

Descriptive logs of cores through the mining zone from the twenty exploratory holes are presented in Figure 2 while Figure 1 shows the location of the holes. For the purpose of this report the term "mining zone" is taken as the interval extending from forty feet above the Mahogany marker to sixty feet below it. Although this demarcation is somewhat arbitrary it is almost certain that this zone will contain the mining height of the future in tracts Ua and Ub.

↑40'
Mahogany Marker
↓60

In the logs, parting planes are planes parallel to the bedding which actually separate the rock core. All observable pre-existing fractures have been noted and their orientation has been described by the angle from the plane transverse to the core axis. Marlstone assaying over 3 gallons/ton in oil content is considered oil shale. Leaner marlstone is described simply as marlstone. These logs will be referred to later in this report.

3 gal/ton

One other notable point is that core recovery in any of the twenty holes was above 90% and averaged 98.0%. This indicates that the cored interval in each hole is composed of fairly competent ground. In Table 1, certain relevant data for the twenty holes are presented. Reference to this table will be made later in this report.

3. LABORATORY TESTS

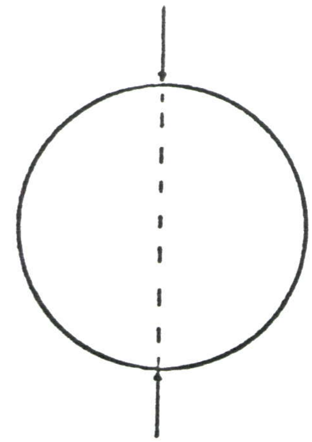
Selected lengths of cores from the thirteen most recent holes, namely P-1 through P-4, X-1 through X-6 and X-9 through X-11 were sent to the U. S. Bureau of Mines in Denver, who carried out a number of laboratory tests on them as part of a joint industry-Bureau cooperative agreement program to test and assemble basic rock mechanics data in oil shale. The results of these tests are presented in this section.

3.1 Types of Tests

The tests carried out by the U. S. Bureau of Mines are listed below with a brief description of each test and its purpose.

a) Unconfined Compressive Test: In this test a cylindrical test-piece of rock is held between the parallel platens of a testing machine on its flat ends and subjected to a monotonically increasing compressive load parallel to the axis of the cylinder until failure occurs. Longitudinal and lateral deformations of the test-piece are measured as the load is increased. The load at failure is used in calculating the unconfined compressive strength of the rock while stress-strain curves derived from the measured load and deformations yield the Young's Modulus and Poisson's Ratio.

b) Brazilian Test: This test involves subjecting a cylindrical test-piece of rock to increasing compressive load along one of its diametral planes as shown in the adjoining sketch until failure occurs by splitting along the plane of loading. The load at failure is used in calculating the maximum tensile stress acting across the plane of failure which is considered as a close, although not exact, estimate of the tensile strength of the rock.



c) Triaxial Test: In this test the curved surface of a cylindrical test-piece is covered by a thin membrane and a compressive load is applied on the flat ends while the test-piece is contained in a chamber of hydraulic fluid maintained at a specified pressure. The end load is increased until failure occurs but because of the confinement offered by the fluid pressure the load carrying capacity of the rock is increased. The triaxial compressive strength of rock is thus higher than the unconfined compressive strength, the magnitude of the increase depending upon the confining pressure.

d) Creep Test: When a test-piece of rock is subjected to a load it undergoes an initial amount of instantaneous deformation. If the applied load is maintained constant at the original level after the initial deformation the test-piece will further deform with time even if the load is not increased. This time-dependent deformation is also called creep deformation, the rate of which depends on the rock type, ambient temperature, the level of the applied load, etc. A knowledge of the time-dependent deformation properties of the rock is necessary for a reliable estimate of the long-term stability of any structure in rock such as mine pillar or a roof strata.

In the creep tests carried out by the U. S. Bureau of Mines a cylindrical test-piece is subjected to a constant uniaxial load, usually 50% or 75% of the unconfined compressive strength, and the deformation is measured at regular intervals over a period of up to 30 days. A plot of the measured deformation against time shows the rate of creep deformation.

e) Specific Gravity Measurement: Specific gravity is determined from the measured weight and volume of the test-piece. This is apparent specific gravity; however, the natural porosity of oil shale is low and the apparent specific gravity is considered to be close to the true specific gravity. For oil shale, specific gravity has been found to be a useful index of other properties such as oil content, compressive strength, etc.

Besides the above tests carried out by the U. S. Bureau of Mines in Denver on cores from the P and X holes, results of tests carried out in 1972 on cores from the four S holes for Gulf Mineral Resources Co. are also available. These latter tests performed by the Colorado School of Mines include the unconfined compressive test, the Brazilian Test and the Direct Tensile Test. The results of these tests have also been included in this report. The Direct Tensile Test involves pulling apart a cylindrical test-piece of rock and measuring the load at failure which enables one to determine the maximum tensile stress withstood by the rock and hence the tensile strength of the rock.

3.2 Results

In presenting the results of the above tests an attempt has been made to derive a total picture for the two tracts. Data from all of the seventeen holes has been considered and processed, wherever necessary.

All processing has been explained. Raw data is not presented mainly because of bulk. However, copies of all such data have been transmitted to the Engineering Manager, White River Shale Project.

It will be found in the following sections that when considering the results of a particular test a small number of results have been rejected because they did not fit the range displayed by the other results. This is an accepted practice in rock mechanics data processing. Rocks being a natural substance inevitably containing inherent flaws, it is impossible to obtain completely reproducible results.

3.2.1 Unconfined Compressive Test

Test-pieces having length:diameter ratios of 1:1, 2:1 and 2.5:1 were used for these tests. It is well known that the unconfined compressive strength of rock varies with the length to diameter ratio of the test-pieces (the higher the ratio the lower the strength value obtained). In order to maintain uniformity all the strength values obtained by the tests have been converted to equivalent strength of 1:1 length:diameter ratio test-piece by the following standard equation:

$$Co = \frac{Cp}{0.778 + 0.222 \times \frac{D}{L}}$$

where, Cp = Strength obtained from test

D/L = Diameter to length ratio of test-piece

Co = Equivalent strength of 1:1 test-piece

The reason for choosing 1:1 ratio as standard is that pillars in the future mine will have approximately that ratio in actuality.

The results are presented in Table 2 which shows the mean values of the compressive strength, Young's Modulus and Poisson's Ratio together with the standard deviation as a percentage of the mean and the number of tests.

Segregation of the results, according to location with respect to the Mahogany Marker, was carried out by considering the range of values obtained from different holes. In doing so 6% of the data from the 165 individual tests had to be rejected because they did not fit the other results for that horizon. As shown in Table 2 the standard deviation for the compressive strengths obtained by this segregation is quite good.

The compressive strength values from Table 2 are plotted in Figure 3 as a histogram over the mining zone, that is, from forty feet above to sixty feet below the Mahogany Marker. Figure 3 shows that the weakest region in the mining zone is from -12.8 to -19.3. The compressive strength of this region is 10,298 PSI ± 13.98%. The strength histogram in Figure 3 can be useful in the future in designing blasting patterns.

In Figure 4 is shown a histogram of oil content over the mining zone. The interval used in this diagram is one foot and the oil content data was obtained by averaging the Fischer Assay results for each horizon from all twenty holes. This is a simple arithmetic average and is not weighted by the density of the rock. Comparison of Figure 3 and Figure 4 indicates that the compressive strengths match very well with the oil contents in accordance with the well known fact that the higher the oil content of oil shale the lower is its strength.

Young's Modulus was derived from the load-deformation (longitudinal) plot obtained from the unconfined compressive tests. The modulus reported is the secant modulus at a load of 20,000 lbs. which is equivalent to a little more than 5,000 PSI. The load-deformation plots were essentially straight lines in this range. The load-deformation (lateral) plot was used for deriving the Poisson's Ratio. Examination of Table 2 will reveal that the Young's Modulus and Poisson's Ratio values follow the general rule that the higher the strength of rock the higher the Young's Modulus and the lower the Poisson's Ratio. However, the standard deviations for both these quantities are rather high.

3.2.2 Brazilian Test

The Brazilian tests on cores from the P and X holes were carried out using test-pieces of length to diameter ratio of 2.5:1 while those on the cores from the S holes were done using test-pieces of length to diameter ratio of approximately 0.67:1. This difference is not expected to affect the results.

The Brazilian test results are shown in Table 3. The intervals of Table 2 used in segregating the compressive test results are used in segregating these results as well. Twelve out of ninety test results were rejected because they did not fit the range of data for that interval. The standard deviations are good. Results of the Brazilian tests will be referred to later in the report.

3.2.3 Triaxial Test

The triaxial tests were carried out using test-pieces of length to diameter ratio of 2.5:1. The results are presented in Table 4 after segregating them according to the intervals in Table 2. Of the 148 test results 9.45% had to be rejected because the values did not fit the range for that interval. In order to make a comparison the unconfined compressive strengths are also shown in Table 4 after converting the values given in Table 2 to equivalent strength of 2.5:1 length/diameter test-pieces using the equation in section 3.2.1.

Table 4 shows that the extent of increase in strength due to the confining pressure varies, the maximum increase being 54% for 500 PSI confining pressure and 94% for 1000 PSI confining pressure. The average increases are 21% and 28% for 500 PSI and 1000 PSI respectively. However, there are some discrepancies in that the strength for 1000 PSI confining pressure is lower than that for 500 PSI confining pressure in a few cases. Also, in one instance, the strength for 500 PSI confining pressure is lower than the unconfined strength. This can be explained by the fact that the triaxial test is very sensitive to inherent flaws in the test-piece. The existence of a thin tar seam can drastically lower the strength and, with the limited number of tests carried out, this affects the average value significantly.

3.2.4 Specific Gravity Measurement

The results of specific gravity measurements are shown in Table 2 alongside the unconfined compressive test results. The standard deviations for the specific gravity measurement are excellent.

3.2.5 Direct Tensile Test

Direct tensile tests were carried out in 1973 on cores from the four S holes. The work was done by the Colorado School of Mines for Gulf Mineral Resources Co. This test is very sensitive to certain conditions which are extremely difficult to control. The tensile strength value of rock obtained by direct tensile tests is not used for any design purposes. Nevertheless, the results are presented in Table 5 for the sake of record.

3.2.6 Creep Test

Because of their time consuming nature the full range of creep tests are not yet complete. The limited amount of results obtained from the Bureau of Mines in Denver is presented in Table 6.

If the time-dependent deformation of rock is plotted against time the general shape of the curve will be as shown in Figure 5. This curve can be divided into three stages as shown in the figure. In the primary stage deformation takes place at a high but diminishing rate. The rate of deformation then settles down to a constant as the secondary stage is reached. Finally the rate accelerates as the tertiary stage is reached until failure takes place. The total amount of deformation in the primary stage, the rate of deformation in the secondary stage and the duration of the secondary stage all depend on the level of the constant load applied and prevalent conditions of temperature, etc. The creep rate may be zero depending upon the level of load in relation to the nature of the rock.

In Table 6 the creep rate shown is the secondary creep rate. The oil content of the test-piece is also given to enable comparisons to be made. These gallons/ton values are the results of Fischer assay on the test-piece alone, not on the one-foot interval containing the test-piece as employed in the general assay work for reserve calculations.

It is difficult to make any interpretation of the results of the limited number of tests carried out so far. However, the expected trend of low grade oil shale showing lower creep rate is borne out to some extent. Two of the tests were carried out under a confining pressure of 500 PSI which results in a lowering of the creep rate as expected. More creep tests, both unconfined and under confining pressures are being carried out.

This is seen in one of the two tests in Table 6 carried out under confined conditions. In one of these tests a confining pressure of 500 PSI lowered the creep rate from 14.05 to 8.23 microinch/inch/day.

Although the pillars in a flat room-and-pillar mine are apparently subjected to a uniaxial load it is well known that the core of the pillar (approximately 50-70% of the volume) is under a small but significant amount of confining pressure offered by the skin of the pillar. The effect of this confining pressure is twofold. First, the load carrying capacity of the core of the pillar is increased as mentioned in section 3.2.3. This means that the pillar factors of safety as computed in Table 8 are conservative estimates. Secondly, the confining pressure lowers the creep rate thereby reducing the deformation of the pillar.

It is, therefore, necessary to carry out some more creep tests under confining pressure with test-pieces of various grades of oil content.

9. NAHCOLITE OCCURRENCES

Nahcolite is a mineral frequently found in oil shale deposits. Chemically it is a bicarbonate of sodium and is readily soluable in water. In the Piceance Creek and Uintah basins nahcolite occurs extensively well below the Mahogany zone while in the Mahogany zone itself nahcolite occurrences are rather erratic.

In tracts Ua and Ub nahcolite has been intercepted in measurable amounts within the mining zone in four out of the twenty exploration holes. This is shown in the lithologic logs of Figure 2. The average thickness of these intercepts is less than two feet and they occur in a zone located from 15 feet to 22 feet below the Mahogany Marker. Summarizing, nahcolite occurrences are likely to be encountered in the mining zone of Ua and Ub in isolated pockets, no more than two feet in thickness and located approximately 20 feet below the marker. The lateral extent of these pockets cannot be predicted but are unlikely to be much greater than their thicknesses.

9.1 Effect of Nahcolite on Pillar Stability

Although nahcolite itself is a mechanically strong mineral, its high solubility in water gives rise to problems whenever it occurs in a mine. It has been found in the Colony Mine in the Piceance Creek basin that whenever nahcolite is exposed it is gradually leached away by atmospheric moisture and mine water until a large cavity is formed. Such a cavity, if formed in a pillar, is likely to endanger the stability of the pillar because concentration of stress will take place around the cavity, resulting in local failure and gradually leading to large-scale failure.

As mentioned earlier nahcolite is expected to occur in the mining zone of Ua and Ub in isolated pockets. As long as such pockets are totally enclosed within the pillar rock they will not cause any problem. However, if appreciable amounts of nahcolite are exposed, the best way to deal with them is to seal off the exposed region to prevent erosion of the nahcolite. The sealing may be done by applying an effective layer of either shotcrete or any other water sealing coating.

10. MINE ROOF AND SUPPORT REQUIREMENT

It has been shown in section 7.3.1 that the roof of the proposed mine is very likely to be located at +20 feet with respect to the Mahogany Marker. This will provide a 4.35 foot thick immediate roof bed having a compressive strength of 28,000 PSI followed by a 5.3 foot thick bed having a compressive strength of 19,000 PSI.

The roof will thus be composed of competent rock. And if no persistent jointing is found to intersect the roof rock such a roof will not require to be supported by systematic bolting in all exposed areas. Areas in producing panels which will be in use for a relatively short period of time are unlikely to require any bolting from the point of view of roof stability although systematic bolting may be a statutory requirement. However, all permanent excavations will have to be supported by bolting.

11. FUTURE WORK

In the preceding pages of this report reference has been made to certain gaps in the rock mechanics data available to date. It is important to collect further data to fill these gaps. This will enable one to make refinements in the designs presented here, thereby gaining greater confidence in them. Such additional work considered necessary is listed in this section.

11.1 Field Work

A thorough survey of all structural features such as joints, parting planes, vugs, etc. in the mining zone should be carried out. Access to the mining zone by a test shaft is necessary for carrying out such a survey. The findings of an underground structural survey will enable one to make a definite decision on the location of the roof and a better assessment of the stability of the roof strata and of the pillars. It may be mentioned that a visual examination of the outcrops at the Hells Hole Canyon revealed some jointing. However, these are of a superficial nature and are in no way indicative of the existence of jointing underground.

Stress measurements should be made in the mining zone to determine the virgin stressfield. It is particularly important to check the existence of any frozen horizontal stress of tectonic origin. Such primitive horizontal stresses are anticipated in a mountainous region where a free face for stress-relief does not exist. It has been pointed out that the Hells Hole Canyon, situated to the east of Ub, may have acted as such a free face. However, the minimum distance between this canyon and the eastern boundary of Ub is approximately one mile. Considering this, it is unlikely that Hells Hole Canyon acted as a free face for the relief of horizontal stresses except, perhaps, for the eastern fringes of Ub.

11.2 Laboratory Tests

The factors of safety for roof stability computed in section 7.3.1 are based on the assumption that the modulus of rupture of the roof strata is two and one-half times its Brazilian strength. A more accurate estimate of the factor of safety can be made if the measured value of the modulus of rupture is used for this purpose. Modulus of rupture tests should, therefore, be carried out on cores from the roof rock. These cores should be taken parallel to the bedding so that the loading during the test simulates the actual loading of the roof strata in the mine.

In section 7.3.2 it has been mentioned that the roof sags tabulated in Table 9 are due to instantaneous elastic deformation of the roof strata.

Further sagging of the roof will take place because of creep deformation of roof strata. In order to assess the stability of the roof with regard to time-dependent sag, it is necessary to conduct some creep tests in bending. These tests need to be carried out on cores of roof rock alone. These cores should also be taken parallel to bedding.

Finally, creep tests in compression now being carried out by the U. S. Bureau of Mines in Denver, are considered adequate. As mentioned earlier Fischer assays should be done on all creep test samples. One or two long-term creep tests extending over a year should also be carried out to check any decrease of the secondary creep rate.

11.3 Analytical Work

The designs presented in this report are based on manual analysis of stresses and deformations. However, certain stress situations in a room-and-pillar mine are best analyzed by numerical methods with the help of the digital computer. For example, the distribution of shear stresses in the pillar and the roof can be accurately determined by The Finite Element Method of analysis. It is recommended that a Finite Element analysis be carried out when all the future data collection outlined in this section is completed. Such an analysis will enable one to verify the stability of the roof and the pillar with respect to shear stresses, further refine the room and pillar designs already made and also predict the effect of mining on the ground surface.

Further, if stress measurements recommended in section 11.1 reveal the existence of horizontal stresses, particularly of a multidirectional nature, an in-plane analysis should also be carried out. This will help one to select the best orientation for the mine and the optimum sequence of mining with respect to the prevalent stressfield.

TABLE 1

Data From Exploration Holes

Hole Number	Collar Elevation	Depth to Mahogany Marker	Cored Interval + Above M.M. - Below M.M.	Core Recovery	True Dip of Mahogany Marker: Based on Data from Holes	Amount of Dip	Direction
P-1	5284.0	745	+681 to -493	99.0%	P-4,E-2,X-6	2°57'32"	S88°36' W
P-2	4990.5	819	+725 to -371	95.1%	E-2,X-3,X-6	2°53'26"	N86°33' W
P-3	5490.0	796	+726 to -424	98.5%	X-6,X-3,E-1	2°36'10"	N74°27' W
P-4	5718.5	703	+651 to -473	99.0%	X-3,P-1,E-1	3° 7'17"	N76° 9' W
X-1	5293.5	1003	+133 to -147	99.6%	E-1,P-1,X-2	1°39'55"	N57°18' W
X-2	4984.0	641	+181 to -125	95.6%	X-6,E-1,X-5	2°39'45"	N57°33' W
X-3	5295.0	470	+149 to -121	99.9%	E-1,X-2,X-4	1°39'55"	N62°45' W
X-4	5242.5	837	+177 to -121	91.5%	X-5,E-1,X-4	2°10'38"	N45°30' W
X-5	5381.5	812	+142 to -124	99.2%	E-3,X-4,X-5	2° 1' 1"	N59°30' W
X-6	5472.0	682	+172 to -124	99.7%	X-6,X-5,E-3	2°29' 3"	N60°42' W
X-9	5427.0	944	+128 to -124	98.8%	P-4,X-6,E-3	2°31'34"	N61°12' W
X-10	5463.5	972	+132 to -121	99.8%	E-3,X-4,S-1	1°51'59"	N50°54' W
X-11	5261.0	866	+176 to -157	97.5%	E-3,X-9,S-1	1°58'37"	N45°27' W
S-1	5242.0	872	+ 55 to -269	99.8%	E-3,X-9,P-3	2° 1'44"	N45°27' W
S-2	5432.0	868	+ 47 to - 96	94.1%	P-3,X-9,S-2	2° 5'27"	N40° W
S-3	5334.0	1039	+ 79 to - 91	99.8%	S-2,X-10,X-11	1°46'21"	N43° W
S-4	5061.0	949	+ 39 to - 88	99.9%	S-2,X-11,X-9	1°45'24"	N32° 6' W
E-1	5292.0	779	+314 to -120	98.2%	X-9,X-11,S-1	1°42'12"	N36° 6' W
E-2	5435.0	504	+154 to -246	99.6%	X-11,P-2,S-1	1°44'22"	N36°11'42" W
E-3	5433.0	842	+410 to - 98	96.0%	X-10,X-11,S-3	1°35'18"	N35°30' W
					X-10,X-1,S-3	1°34' 6"	N39°30' W
					S-3,X-1,S-4	1°38' 4"	N39°21' W
					S-3,P-2,S-4	1°38'50"	N27°24' W
					X-11,S-3,P-2	1°45'57"	N31° 0'48" W

NOTE:

Average Depth to Mahogany Marker:
1. In Ua 926.0
2. In Ub 710.0

Average Core Recovery: 98.0%

NOTE:

Average Dip of Mahogany Marker:
1. In Ua 1°32'33" or 2.69%
2. In Ub 2°26'56" or 4.28%

TABLE 2

RESULTS OF UNCONFINED COMPRESSIVE TESTS

LOCATION OF INTERVAL WITH RESPECT TO MAHOGANY MARKER + ABOVE (IN FEET) - BELOW (IN FEET)	UNCONFINED COMPRESSIVE STRENGTH CONVERTED TO 1:1 RATIO			YOUNG'S MODULUS			POISSON'S RATIO			SPECIFIC GRAVITY		
	Mean Value PSI	Per Cent Standard Deviation	No. Of Tests	Mean Value PSI X 10^6	Per Cent Standard Deviation	No. Of Tests	Mean Value	Per Cent Standard Deviation	No. Of Tests	Mean Value GMS/cc	Per Cent Standard Deviation	No. Of Tests
+46.9 to +37.61	19031	13.28	7	2.64	25.81	7	0.28	27.08	7	2.269	8.45	6
+37.61 to +30.9	34153	18.58	6	3.73	13.81	6	0.162	21.22	6	2.448	1.21	6
+30.8 to +25.61	18950	15.07	9	4.22	27.88	9	0.17	37.77	9	2.298	3.21	6
+25.61 to +21.25	27983	19.49	5	3.40	24.51	5	0.19	21.08	5	2.484	1.24	5
+21.25 to +15.18 6.07	18145	9.20	12	2.50	17.65	12	0.217	46.8	12	2.399	3.03	7
+15.18 to + 8.4 6.78	12721	20.75	12	1.45	30.97	12	0.303	17.88	11	2.107	6.52	8
+ 8.4 to 0 8.4	21120	19.46	12	2.58	27.07	12	0.287	27.07	12	2.338	2.63	11
0 to - 4.02 4.02	25957	15.20	9	3.38	26.32	9	0.269	15.12	9	2.419	1.6	8
- 4.02 to -12.8 8.78	13514	11.87	21	1.34	25.75	21	0.322	16.18	21	2.108	6.27	20
-12.8 to -19.30 6.5	10298	13.58	6	0.54	54.36	5	0.469	24.18	6	1.714	3.12	4
-19.30 to -30.8 11.5	11228	16.07	16	0.94	36.82	15	0.42	30.67	15	1.968	6.04	16
-30.8 to -39.8 9.0	17062	10.12	17	2.15	20.62	17	0.265	28.63	15	2.269	2.56	15
-39.8 to -42.8	18308	17.54	6	2.165	31.43	6	0.27	29.34	6	2.266	5.24	5
-42.8 to -49.8	13998	17.85	9	1.55	35.99	9	0.33	28.35	9	2.198	4.81	8
-49.8 to -57.5	17479	14.01	13	2.85	23.44	13	0.23	34.44	13	2.305	1.23	9
-57.5 to -60.8	24412	17.60	6	3.23	27.32	6	0.23	19.11	6	2.358	3.47	6

TABLE 3

BRAZILIAN TEST RESULTS

Location of Interval With Respect To Mahogany Marker + Above (in feet) - Below (in feet)	Brazilian Strength Mean Value PSI	Per Cent Standard Deviation	Number of Tests
+46.9 to +37.6	860	24.3	3
+37.6 to +30.9	1255	14.3	4
+30.9 to +25.61	1108	22.4	5
+25.61 to +21.25	1072	17.2	3
+21.25 to +15.18	1079	8.6	6
+15.18 to + 8.4	1096	11.7	4
+ 8.4 to 0	1267	14.9	5
0 to - 4.02	1276	20.2	6
- 4.02 to -12.8	949	16.0	9
-12.8 to -19.30	1174	5.5	4
-19.3 to -30.8	1170	11.6	9
-30.8 to -39.8	1147	15.7	7
-39.8 to -42.8	903		2
-42.8 to -49.8	1104	9.7	5
-49.8 to -57.5	1223	17.4	6
-57.5 to -60.8	1172	16.8	4

TABLE 4

RESULTS OF TRIAXIAL TESTS

Location of Interval With Respect To Mahogany Marker +Above -Below	Unconfined Compressive Strength Converted to 2.5:1 L/D Ratio	Triaxial Strength at 500 PSI Confining Pressure			Triaxial Strength at 1000 PSI Confining Pressure		
		Mean Value PSI	Per Cent Standard Deviation	No. Of Tests	Mean Value PSI	Per Cent Standard Deviation	No. Of Tests
+46.9 to +37.61	16496	17249		2	20569		2
+37.61 to +30.9	29603	36952		1	35046		2
+30.9 to +25.61	16425	21334	11.22	3	20789		2
+25.61 to +21.25	24256	37387		2	30066		2
+21.25 to +15.18	15728	16021	27.19	5	16992	18.51	5
+15.18 to + 8.4	11027	11899	24.57	5	14789	25.68	6
+ 8.4 to 0	18307	24323		2	26705		2
0 to - 4.02	22500	24680	9.69	7	27498	13.32	4
- 4.02 to -12.8	11714	11941	7.51	6	13136	8.97	7
-12.8 to -19.3	8926	13599	27.08	5	17309	16.36	3
-19.3 to -30.8	9732	12360	29.26	7	13669	25.70	5
-30.8 to -39.8	14789	15670	15.48	5	15958	12.33	4
-39.8 to -42.8	15869	16160		2	17720	12.07	3
-42.8 to -49.8	12133	17476	29.0	7	19125	23.79	6
-49.8 to -57.5	15151	16888		1	16511		2
-57.5 to -60.8	21160	18350	22.29	5	23487	22.26	4

TABLE 5

RESULTS OF DIRECT TENSILE TESTS

Hole Number	Location With Respect To Mahogany Marker	Tensile Strength PSI
S-1	+42.15	100
S-1	+36.65	22
S-1	-51.05	515
S-2	+36.1	71
S-2	+27.35	537
S-2	-43.95	114
S-3	+35.75	57
S-3	+25.1	585
S-3	+14.95	122
S-3	- 4.45	143
S-3	-26.45	857
S-3	-52.35	214
S-3	-63.05	257
S-4	+33.8	72
S-4	+28.4	287
S-4	+18.05	230
S-4	+ 9.1	857
S-4	-22.2	29
S-4	-56.9	430
S-4	-65.5	337

TABLE 6

Results of Creep Tests

Hole Number	Location with Respect to Mahogany Marker	Oil Content from Fischer Assay Gal./Ton	Level of Load as Fraction of Estimated Unconfined Compressive Strength	Creep Rate in Microinch/inch/day
P-1	+39.1	23.77	0.5	55.0
P-1	-21.9	15.99	0.5	33.9
P-2	+44.8	12.60	0.5	15.7
P-2	+44.8	13.08	0.5	30.0
P-3	+23.3	3.15	0.5	0
P-3	-66.7	70.50	0.5	14.05
P-3	-66.7	70.50	0.5	8.23*
P-4	+ 9.3	38.31	0.5	37.0
P-4	-20.7	17.92	0.5	14.0
X-1	+14.1		0.33	10.6
X-10	-28.3		0.5	23*

*These two tests were carried out under confining pressure of 500 PSI.

TABLE 7

Compressive Strengths of the Mahogany Zone in Piceance Creek Basin

Distance from Mahogany marker, feet	Bed designation1/	Number of specimens tested	Compressive strength, p.s.i.	Standard deviation, per cent
31.6 above	Roof	2	15,380	-
21.5-26.8 above	do.	2	14,890	-
20 above	do.	3	12,430	1.0
18.5 above	A	1	17,100	-
10 above	B	2	15,000	-
4.2 above	C	1	17,100	-
2.5 below	D	1	19,000	-
4.5 below	D	2	12,650	-
7 below	D	2	11,730	-
10 below	D	1	10,700	-
12.5 below	E	1	12,520	-
14 below	E	3	8,280	5.0
14.6 below	E	5	7,350	2.6
15.5 below	F	3	11,910	.4
17 below	F	1	9,190	-
17.5 below	F	1	12,030	-
18.5 below	F	8	8,160	3.4
20 below	G	1	14,480	-
20.5 below	G	1	14,470	-
23 below	G	2	10,250	-
23 below	G	2	8,600	-
26.6 below	G	2	14,090	-
27 below	G	3	12,960	7.2
31 below	H	3	8,560	5.7
33 below	H	1	13,600	-
37 below	H	1	15,390	-
39.5 below	H	2	17,280	-
46.5 below	I	2	12,700	-

1/ In accordance with their oil content and caking tendency, the mineable oil-shale beds were grouped into 10-lettered beds.

Note: This table is quoted from U. S. Bureau of Mines, Report of Investigations No. 5089, 1954 by R. H. Merrill.

TABLE 8

EXTRACTION RATIOS AND PILLAR FACTORS OF SAFETY

Room Width Feet	Pillar Dimensions Feet	Extraction Ratio	Maximum Pillar Stress in PSI For Depth 1040	For Depth 926	For Depth 845	For Depth 810	Pillar Factor of Safety For Depth 1040	For Depth 926	For Depth 845	For Depth 810
55	45.0 X 45.0	0.7975	5368	4775	4360	4178	1.68	1.88	2.06	2.15
55	50.0 X 50.0	0.7732	4793	4264	3893	3730	1.88	2.11	2.31	2.41
55	55.0 X 55.0	0.750	4348	3868	3532	3384	2.07	2.33	2.55	2.66
55	60.0 X 60.0	0.7278	3993	3553	3244	3108	2.25	2.53	2.77	2.90
55	65.0 X 65.0	0.7066	3705	3296	3009	2883	2.43	2.73	2.99	3.12
55	50.0 X 70.0	0.7333	4076	3626	3311	3172	2.21	2.48	2.72	2.84
55	50.0 X 75.0	0.7253	3957	3520	3214	3080	2.27	2.56	2.80	2.92
55	55.0 X 70.0	0.720	3882	3454	3154	3021	2.32	2.61	2.85	2.98
55	55.0 X 75.0	0.7115	3768	3352	3061	2932	2.39	2.68	2.94	3.07
57.5	50.0 X 50.0	0.7837	5025	4471	4082	3911	1.79	2.01	2.20	2.30
57.5	55.0 X 55.0	0.7610	4548	4046	3695	3540	1.98	2.22	2.44	2.54
57.5	57.5 X 57.5	0.750	4348	3868	3532	3384	2.07	2.33	2.55	2.66
57.5	60.0 X 60.0	0.7392	4168	3708	3386	3244	2.16	2.43	2.66	2.77
57.5	50.0 X 70.0	0.7543	4424	3936	3594	3443	2.03	2.29	2.50	2.61
57.5	50.0 X 75.0	0.7367	4128	3673	3354	3213	2.18	2.45	2.68	2.80
57.5	55.0 X 70.0	0.7316	4050	3603	3290	3152	2.22	2.50	2.74	2.86
57.5	55.0 X 75.0	0.7233	3928	3495	3191	3057	2.29	2.58	2.82	2.94
60	55.0 X 55.0	0.7713	4753	4228	3861	3699	1.89	2.13	2.33	2.43
60	60.0 X 60.0	0.750	4348	3868	3532	3384	2.07	2.33	2.55	2.66
60	65.0 X 65.0	0.7296	4020	3576	3266	3129	2.24	2.52	2.76	2.88
60	50.0 X 70.0	0.7552	4440	3950	3607	3456	2.03	2.28	2.50	2.60
60	50.0 X 75.0	0.7475	4305	3830	3497	3350	2.09	2.35	2.57	2.69
60	55.0 X 70.0	0.7425	4221	3755	3429	3285	2.13	2.40	2.62	2.74
60	55.0 X 75.0	0.7343	4091	3639	3323	3184	2.20	2.47	2.71	2.83
60	60.0 X 75.0	0.7222	3913	3481	3179	3045	2.30	2.56	2.83	2.96

TABLE 9

Hole No.	Location of 3" Thick Tar-Saturated Sandstone Layer With Respect To Mahogany Marker + above / - below
X- 1	+13.8
X- 2	+14.65
X- 3	+13.32
X- 4	+13.6
X- 5	+12.1 (?)
X- 6	+13.9
X-10	+15.15
X-11	+14.7
P- 1	+13.95
P- 2	+16.2 (?)
P- 3	+14.17
P- 4	+13.22

Note: The locations for X-5 and P-2 are doubtful.

TABLE 10

Reserves Based on Fixed Roof Elevation

Mining Height* Roof	Floor	Ua and Ub Barrels x 10^3	Gal./Ton	Ua Barrels x 10^3	Gal./Ton	Ub Barrels x 10^3	Gal./Ton
+15	-35	974,552	27.24	489,617	27.41	484,935	27.08
+21	-35	1,056,678	26.17	535,606	26.61	521,072	25.74
+15	Float	1,218,346	26.41	592,585	26.60	625,761	26.24
+20	Float	1,287,736	25.78	631,777	26.16	655,959	25.43

*Positive and negative signs indicate above and below the Mahogany Marker.

TABLE 11

Roof Factor of Safety and Roof Sag

Room Width Feet	Maximum Tensile Stress PSI	Factor of Safety	Maximum Deflection or Sag Inch
55	374	7.17	0.0574
57.5	409	6.55	0.0686
60	445	6.02	0.0813

LOGS OF EXPLORATORY HOLES FOR THE MINING ZONE

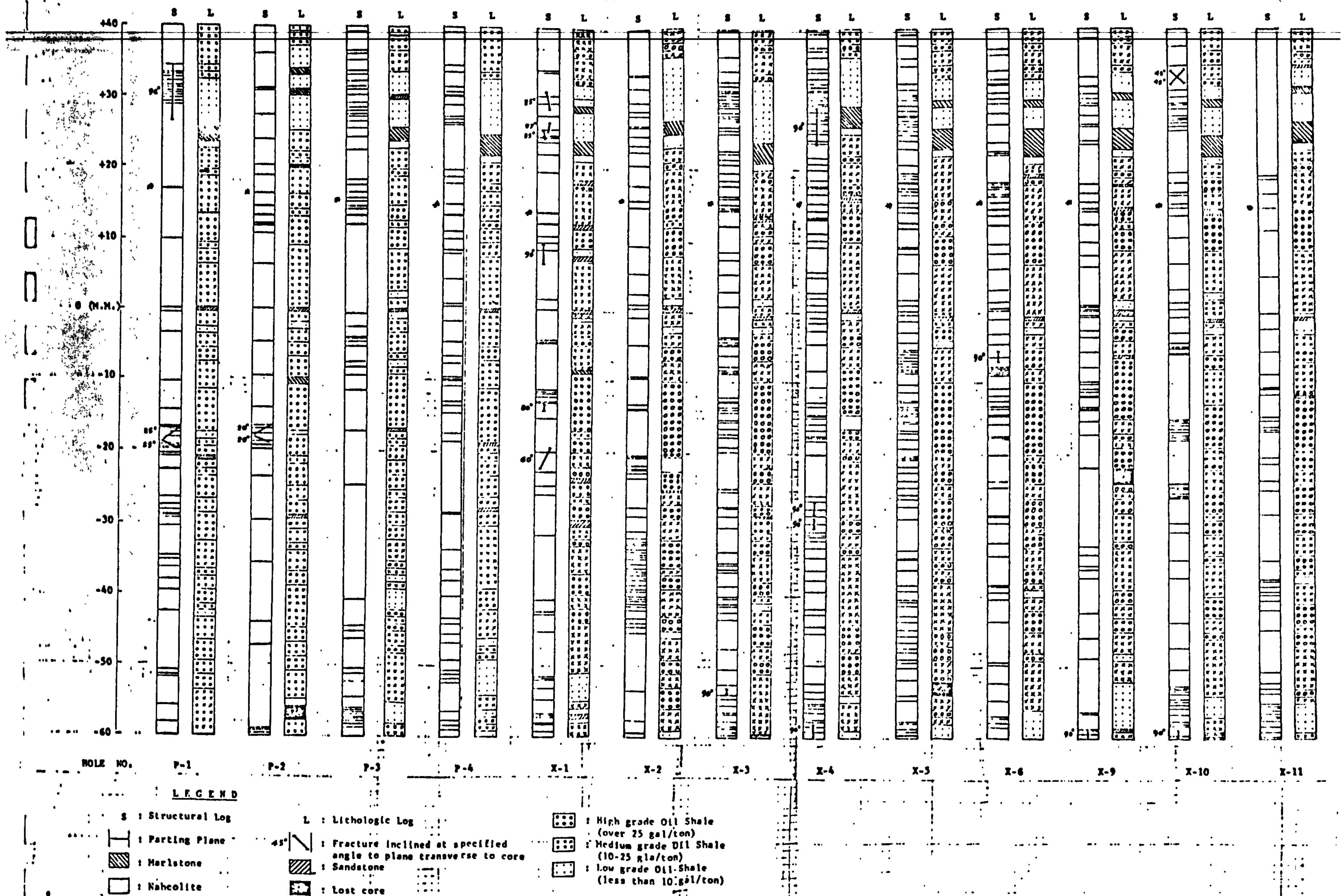
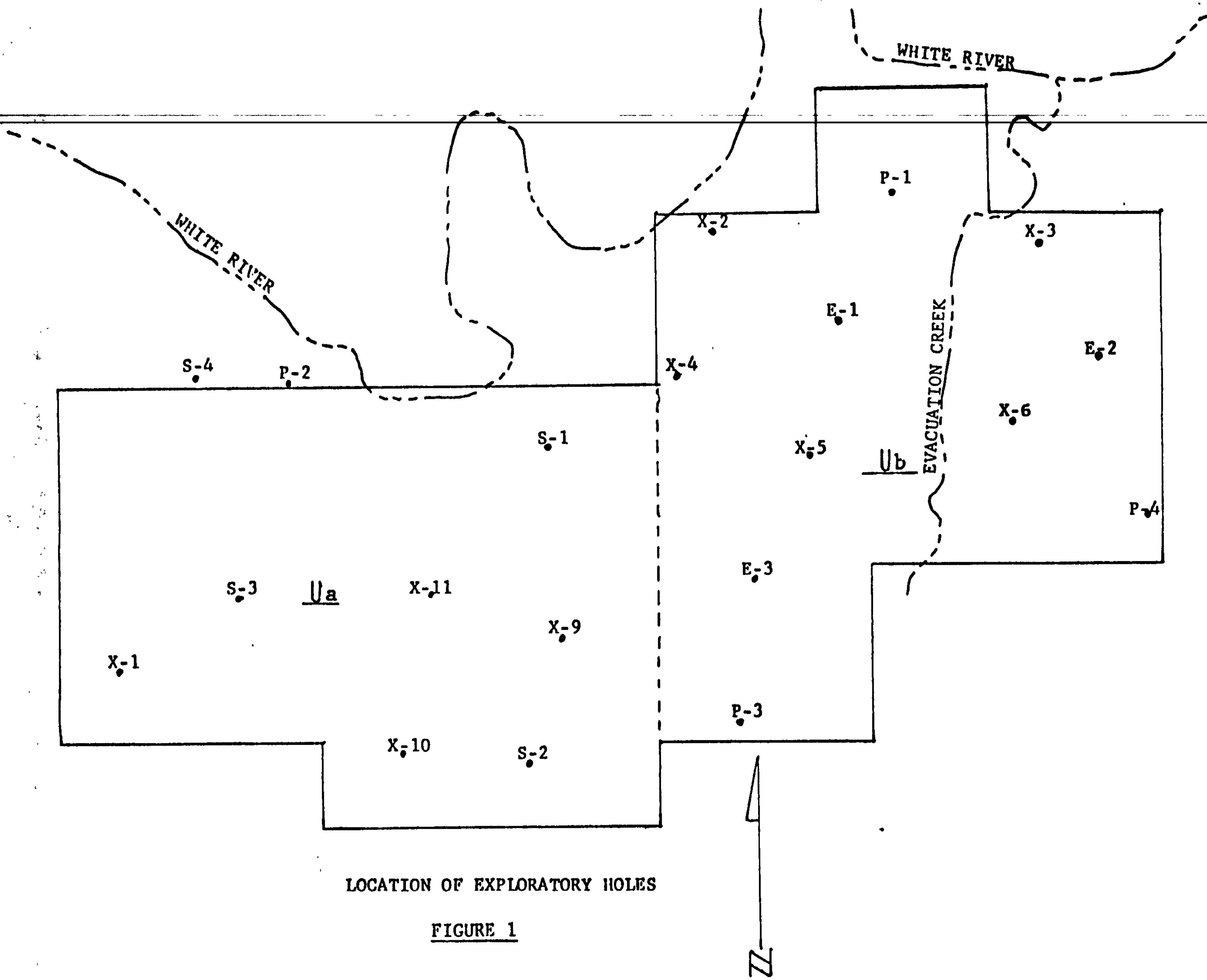
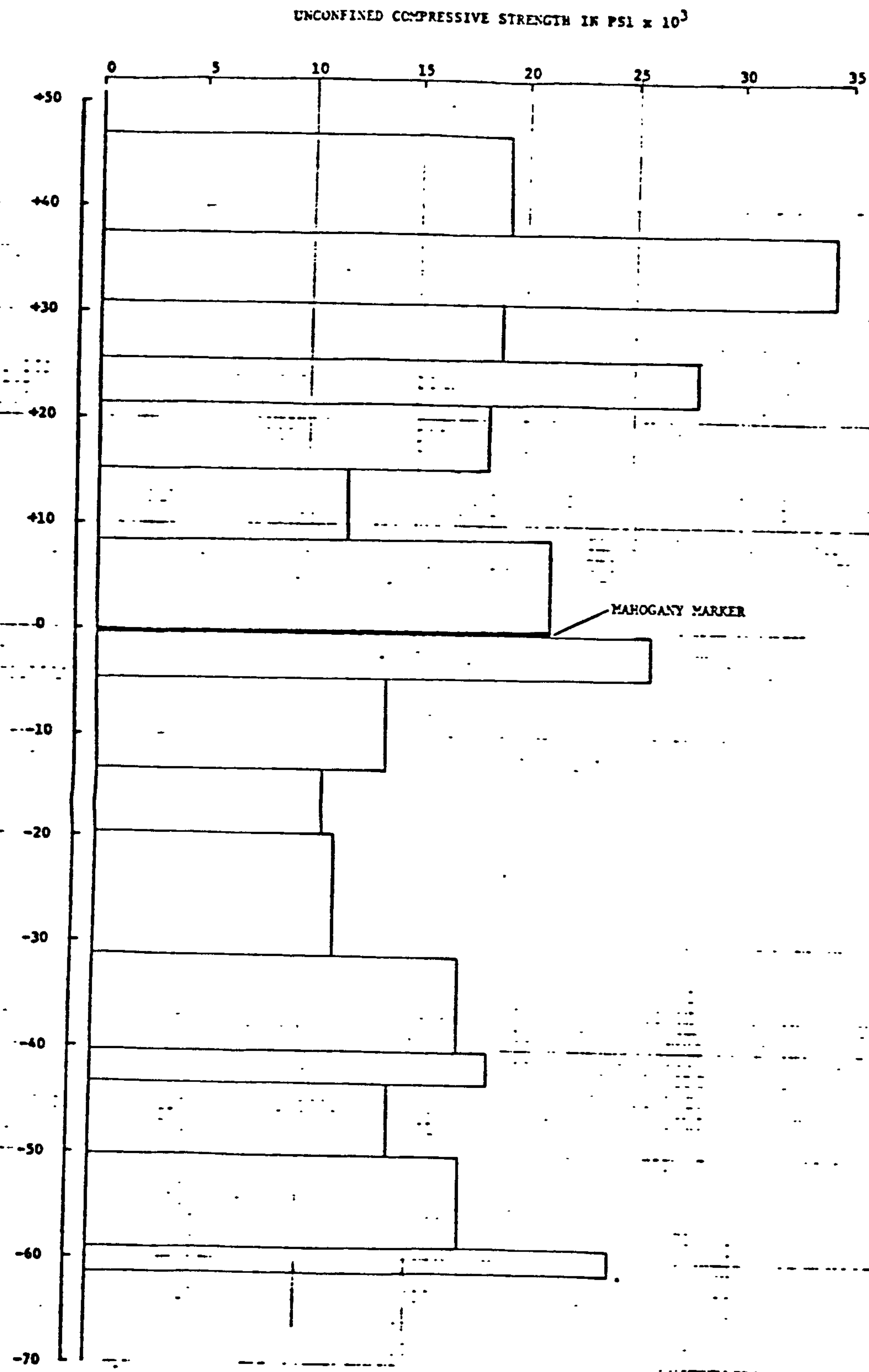


FIGURE 2



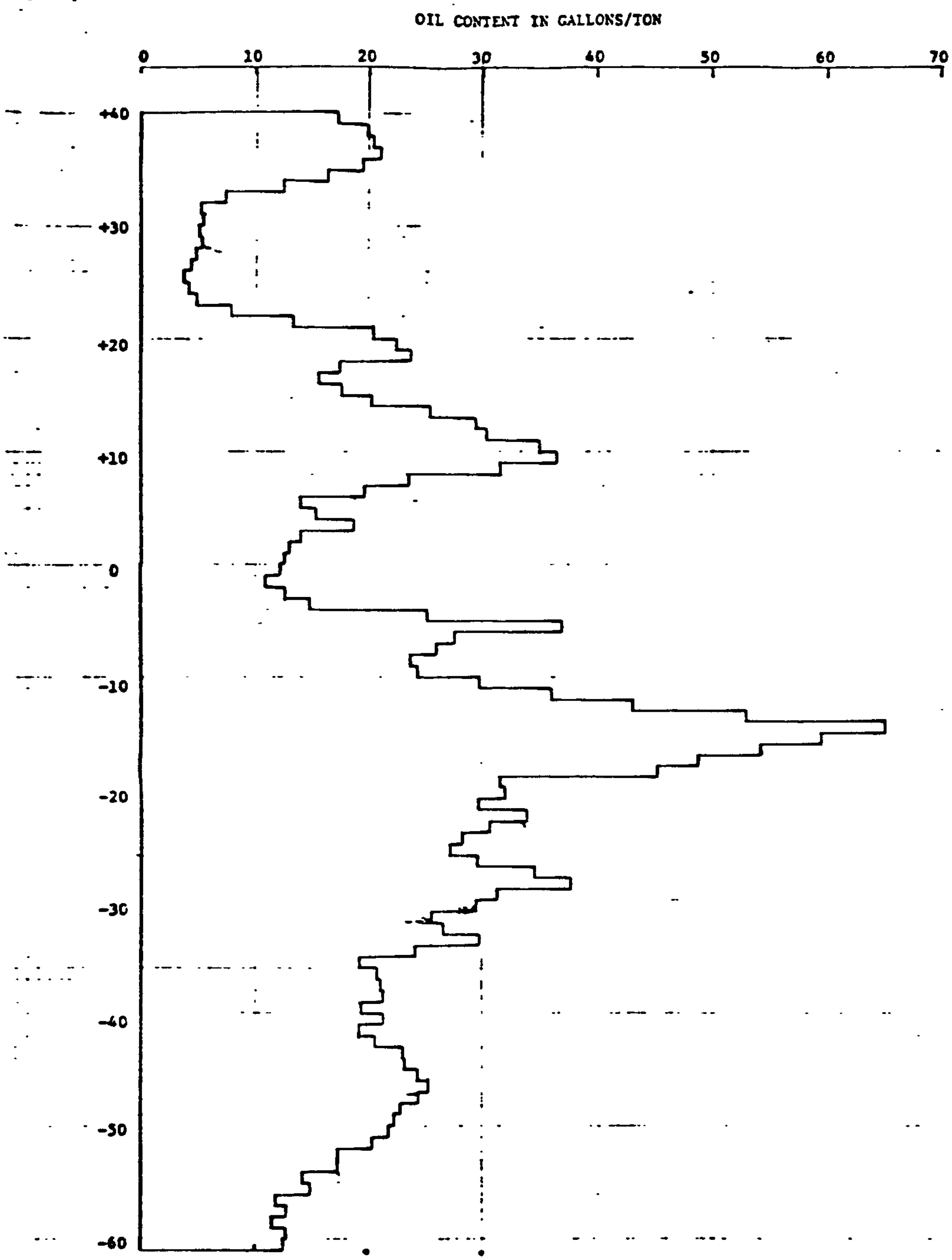
LOCATION OF EXPLORATORY HOLES

FIGURE 1



Histogram of Compressive Strength for Mining Zone

FIGURE 3



Composite Grade Histogram for mining zone by averaging assay data from 20 holes for each horizon

FIGURE 4

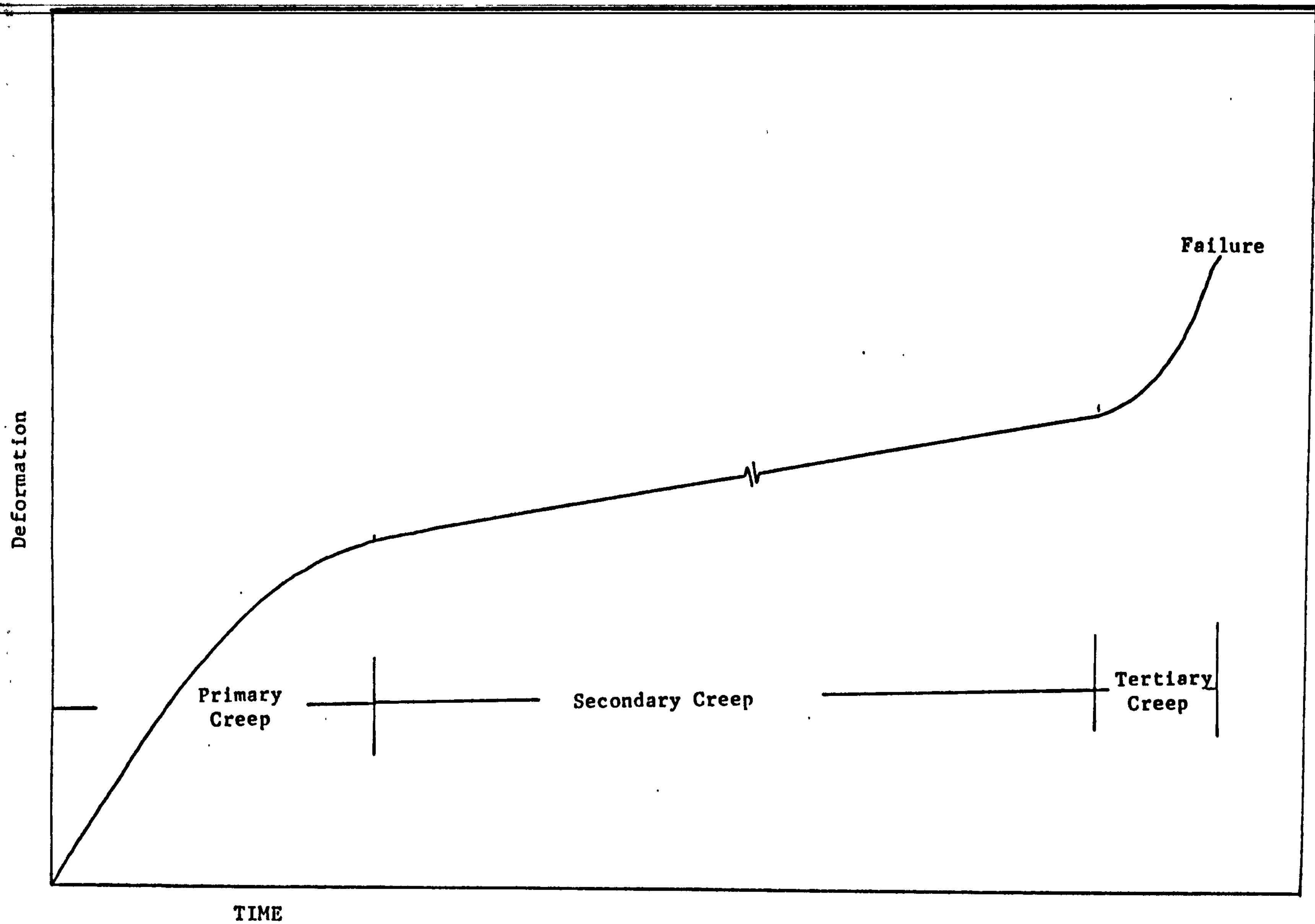


FIGURE 5

Generalized Curve Showing Creep Deformation of Rock

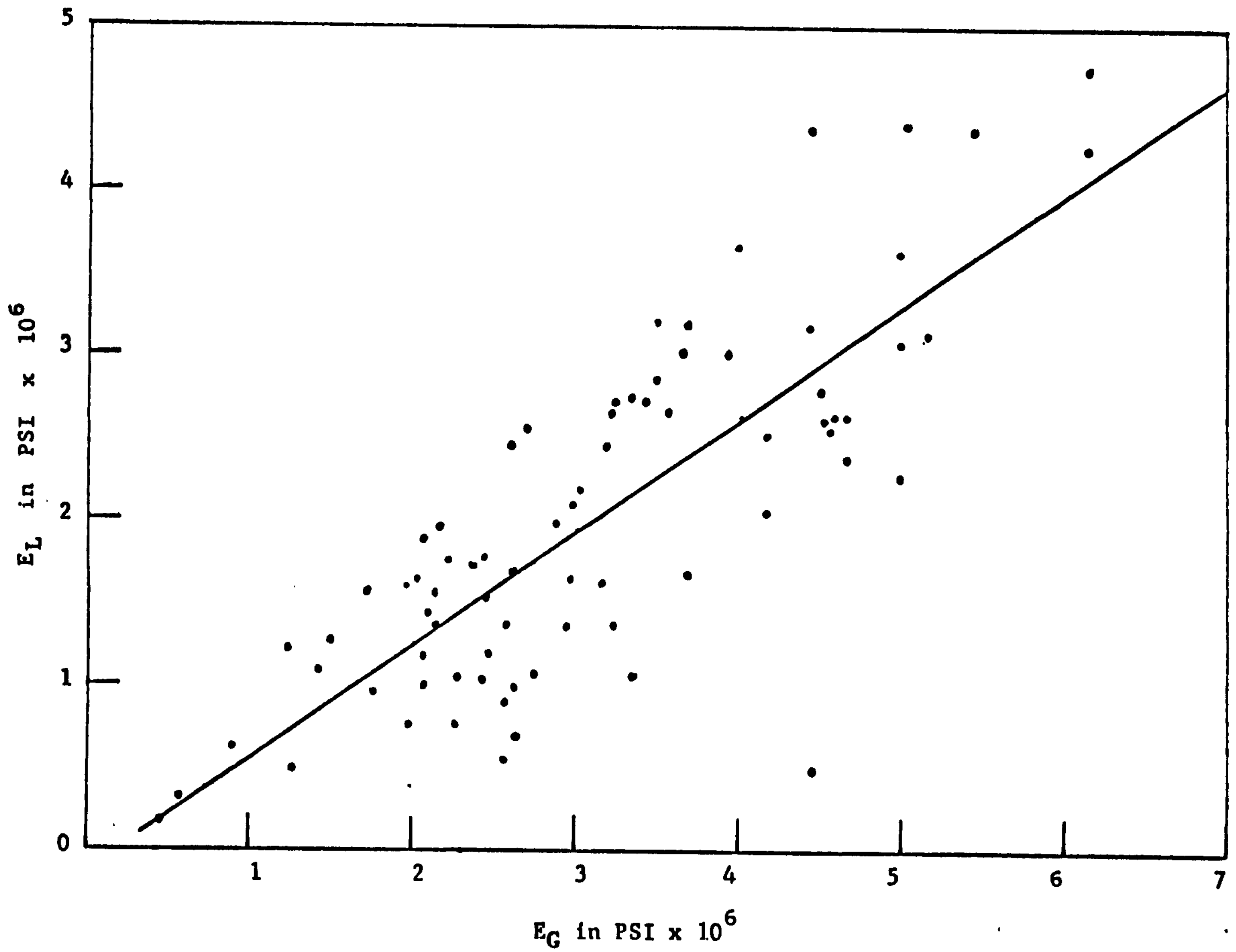


FIGURE 6

Plot of Laboratory Determined Young's Modulus (E_L)
Against Young's Modulus Derived from Geophysical Logs (E_G)

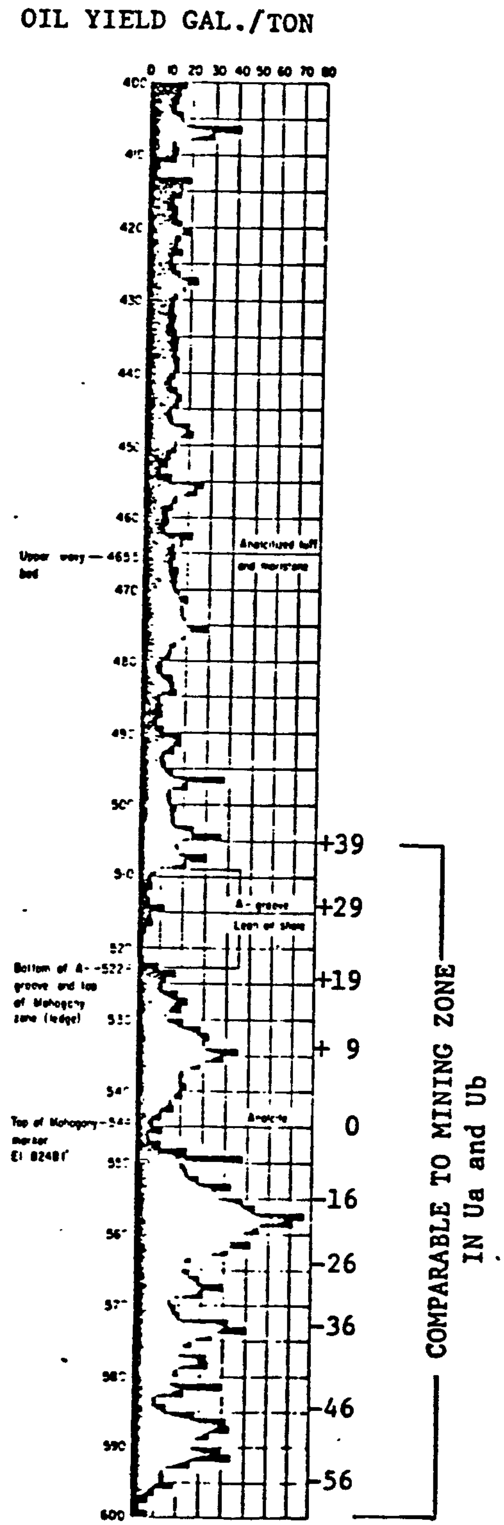


FIG. 7

OIL CONTENT HISTOGRAM OF MAHOGANY ZONE IN PICEANCE CREEK BASIN (ANVIL POINTS)

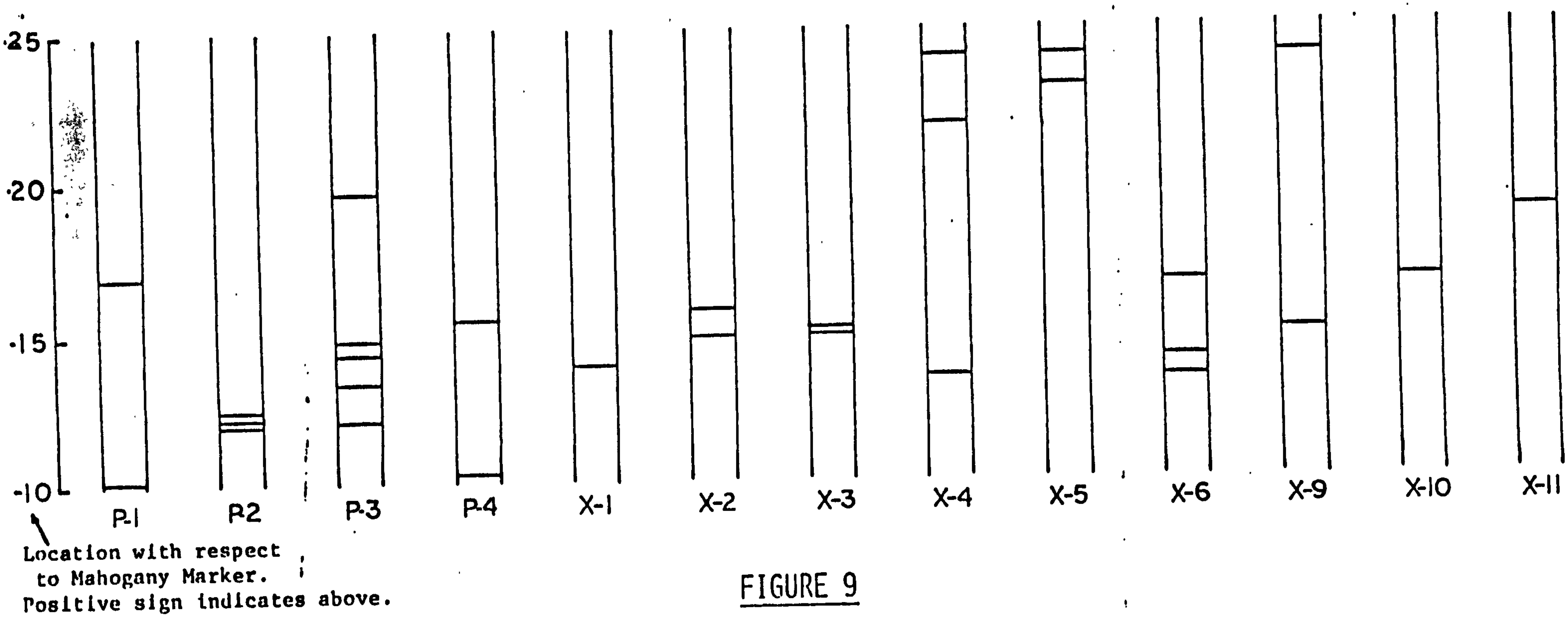


FIGURE 9

LOCATION OF TRUE PARTING PLANES IN ROOF ZONE

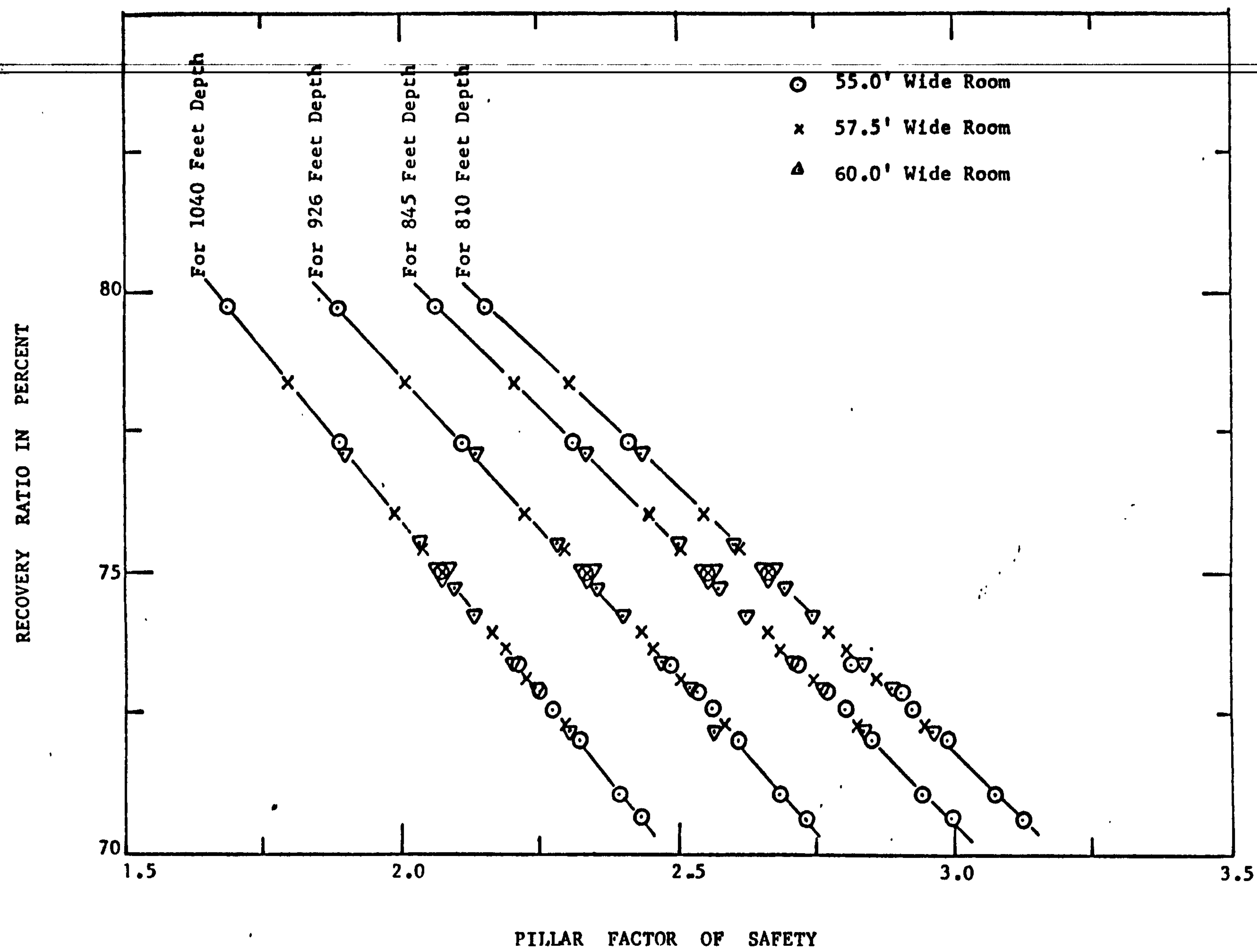

RECOVERY RATIO IN PERCENT
70
75
80
1.5
2.0
2.5
3.0
3.5
PILLAR FACTOR OF SAFETY
For 1040 Feet Depth
For 926 Feet Depth
For 845 Feet Depth
For 810 Feet Depth
55.0' Wide Room
57.5' Wide Room
60.0' Wide Room

FIGURE 8